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MINING METHODS AND COSTS AT THE ENGELS MINE, PLUMAS COUNTY, CALIF.

BY

W. I. NELSON

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MINING METHODS AND COSTS AT THE ENGELS MINE, PLUMAS COUNTY, CALIF.[1]

By W. I. Nelson[2]

INTRODUCTION

This paper describing the mining practices at the Engels mine is one of a series being prepared by the United States Bureau of Mines on mining practices, methods, and costs in the various mining districts of the United States.

The Engels Copper Mining Co. operates two mines about 3 miles apart in Plumas County, Calif. Both mines send ore to the same mill, and the methods of the larger, or Engels mine, are described in this paper. Between 800 and 1,000 tons of 1-3/4 to 2-1/2 per cent copper ore are mined daily from this mine by a shrinkage method.

Connection is made with the Western Pacific Railroad at Paxton by the Indian Valley Railroad, which is 22 miles long and operates a daily mixed train to the mine.

ACKNOWLEDGMENTS

The author is indebted to Mr. O. H. Bru, treasurer of the Engels Copper Mining Co., for the early history of the mine. Some information has been taken from articles by Mr W. R. Lindsay, former superintendent, and by the author.[3] [4] [5]

HISTORY

Copper ore was discovered in the Plumas County district in 1865. A small smelter ran intermittently for four years and was then abandoned as a losing investment.

The Engels mine was located in the late seventies by Henry Engels, and he and his two sons devoted their entire time to the development of the property. The nearest railroad connection at that time was at Reno, Nev., which could be reached over 150 miles of mountain road. Persistent work uncovered considerable ore, some of the richest of which was hauled to Reno and shipped to Swansea, Wales, by way of San Francisco.

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[1] - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6260 "

[2] - One of the consulting engineers, U. S. Bureau of Mines.

[3] - Lindsay, W. R., Construction of Ore Transfer Raise of Engels Copper Mining Co. Eng. and Min. Jour., vol. 120, No. 6, Aug. 8, 1925; also Electric Blasting in the Engels Mine. Eng. and Min. Jour., vol. 120, No. 15, Oct. 10, 1925.

[4] - Nelson, W. I., Removing Vertical Pillars by Diamond Drill Methods. Eng. and Min. Jour., vol. 127, No. 16, April 20, 1929.

[5] - Nelson, W. I., The Application of Multiple Graded Series in Parallel Connections at the Engels Mine. The Explosives Engineer, May, 1929.

In 1901 the Engels Copper Mining Co. was organized. It was not until the reorganization of the company in 1914, however, that operations commenced on a substantially larger scale. At this time a mill with a daily capacity of 150 tons was built and shipments of concentrate were made in addition to those of ore. The concentrate and ore were transported to the bottom of a 1,000-foot hill by an aerial tram and thence by truck to the Western Pacific Railroad at Keddie, 28 miles away. During the dry season three trucks averaged two round trips and transported 20 tons every 24 hours. Transportation difficulties during the winter months led to the construction of the Indian Valley Railroad, which was completed early in 1917. In order to further increase tonnage, the Superior mill, having a crusher capacity of 1,500 tons per day, and a 2-mile aerial tram from No. 6 level were built and completed in the early part of 1918. In 1924 the electric haulage system through No. 10 adit was put in operation and the aerial tram discarded. Since this time the production from the Engels mine has been gradually increased from 800 to 1,000 tons per day. The total production of the mine to date has been 3,335,870 tons of ore.

GEOLOGY

The ore in the Engels mine occurs in shear zones in diorite and altered diorite. The ore bodies vary in length; they have a general strike of N. 60 E. and a dip of 80° to the west. The main ore body consists of three shear zones, two of which are nearly parallel; the third separates from the other two by raking more strongly toward the north. Two other lodes containing commercial ore occur, in one of which the ore is in the form of lenses.

The principal copper minerals are bornite and chalcopyrite, with some secondary chalcocite occurring in the upper levels. The mineralizing solutions evidently followed the channels made by the shearing in the altered diorite; the minerals occur as interlacing networks of small veins in some places and as bunches of ore and disseminations through the rock in other places. The grade of the ore is dependent upon the intensity and the width of the shearing which allowed the mineralizing solutions to permeate through the rock. The proportion of bornite to chalcopyrite is about 2 to 3, and this ratio remains fairly constant with depth; if any change occurs it appears to be an increase in bornite. Above the fourth level much of the ore mined assayed 4 per cent copper, probably due to chalcocite enrichment. Below the fourth level the ore has averaged from 1-3/4 to 2-1/2 per cent copper.

The main ore body had a maximum width of 100 feet and a length of 830 feet. Croppings of carbonates were evidently prominent over the main ore body. The surface rock over the other ore zones, however, was disintegrated and leached to such an extent that very little carbonate was present.

The diorite and altered diorite are very firm, tough rocks which stand well in unsupported walls. There are systems of both vertical and nearly flat cleavage planes and slips which rarely interfere but usually help in the shrinkage stope method. The ground does not require timber for support except occasionally in places where faults cross a drift.

METHODS OF PROSPECTING AND EXPLORATION

The shear zones are usually persistent but sometimes they vary slightly in their downward course. When a level is opened up, drifts are run to the projected position of the ore body and either strike the ore zone or are so near to it that only short crosscuts are required. The occurrence of barren sections in the shear zones has necessitated the use of the diamond drill for exploratory work. The diamond drill has also been employed to prospect for other ore zones and to determine the grade and location of the known ore zones at depth. The holes are put in on contract and are drilled at a base price of $2.45 per foot for holes inclined less than 15° from the horizontal and less than 400 feet long. An increase of 15 cents per foot is made for each 100 feet over the first 400 feet. Holes run at an angle steeper than 15° have a base rate of $2.70 per foot for the first 400 feet and increase 15 cents per foot for each 100 feet thereafter. Up to the present time a total of 75,378 feet of prospect holes has been drilled.

Electrical prospecting was carried on over a considerable portion of the surface near the Engels mine, but this work did not indicate any ore bodies. On account of the occurrence of magnetite with the ore a magnetic survey was also made on the surface, and the results showed weak mineralization at several places. The indicated mineralized areas, however, did not develop into commercial ore bodies.

METHODS OF SAMPLING AND ESTIMATION OF TONNAGES AND VALUES

In development work a pick sample is taken from the face and also a grab sample from the broken rock after each round is blasted.

In stoping, the drill cuttings from two or three holes each day are caught in a powder box and sent to the assay office. The assay result furnishes a guide to the values in the rock being broken. A small grab sample is taken from each car when drawing ore from the chutes; this serves as a check on the grade of the ore from the stopes. The car samples usually run higher than the mill head.

The amount of ore broken is calculated by means of monthly transit surveys of the stopes and of the development workings. The tonnage of ore crushed is determined by a weightometer at the mill. Twelve cubic feet in place is taken as 1 ton of ore, and 18 cubic feet as the tonnage factor for broken ore.

The amount of ore in reserve is calculated from the height and from the average cross section and assay of the area stoped above and below each block. No systematic sampling of ore reserves is done.

METHODS OF DEVELOPMENT AND MINING

General Development

The ore bodies have been developed by means of adits and vertical shafts as shown in Figure 1. The following historical sketch illustrates the general method of development of the mine.

In the upper portion of the mine four adits were successively driven into and along the ore bodies at various elevations. Crosscuts were run and shrinkage stopes started as soon as the development work permitted. The fourth level was driven about 300 feet below the highest cropping of ore. On this level a winze was sunk in the ore, and the so-called shaft level was started 100 feet below. The ore from development and stoping operations was hoisted to the fourth level and trammed to the Engels mill. The sixth-level adit, 2,570 feet long, was driven 250 feet below the shaft level. Raises were run from this level to the shaft level for ventilation, and stoping was started. All of the sixth level ore was transported through No. 6 adit and over an aerial tram to the Superior mill. Figure 2 is a plan of the sixth level, which is a typical haulage level.

No pillars had been left in any of the stopes above the shaft level, and stoping was started according to the same plan on the sixth level. Access to the stopes was by means of cribbed manways. After the stopes had reached a height of about 100 feet these manways caused trouble by leaning over, due to the uneven withdrawal of ore. In addition, the walls of the stope had a tendency to cave when the broken ore was withdrawn upon completion of back stoping. To overcome these difficulties, pillars 20 feet wide were left from foot to hanging wall at intervals of 90 to 120 feet for the remaining distance between the back of the stope and the shaft level.

A raise was run from the sixth level to connect with No. 2 shaft, which was sunk 150 feet from the surface. This shaft was continued downward and from it the seventh level was opened 200 feet below the sixth. Stopes were started, and the ore was hoisted in 2-ton skips to the sixth level. Manway raises with crosscuts every 35 feet were driven in alternate pillars to afford access to stopes for men and supplies.

The No. 10 adit was driven 7,735 feet to intersect the ore body 490 feet below the seventh level, and an electric haulage system was built to the mill. A raise and an ore pass were driven from the tenth level to connect with the seventh (fig. 3). When these were completed a cage was installed in the manway raise. The eighth, ninth, and tenth levels are 168 feet apart vertically. The ore dumped into the ore pass at the several levels is drawn into cars at the tenth level and hauled to the mill.

Shafts and Winzes

Three-compartment shafts are excavated 7 by 17 feet in the clear. Timbers are 8 by 8 inch and are framed as shown in Figure 4. The two hoisting compartments are each 4 feet 10-1/2 inches by 5 feet, and the manway compartment 4 by 5 feet. The sets are placed on 6-foot centers. The shaft timbering without any lagging outside the sets but with one compartment lagged with 3 by 12 inch plank requires 102 board feet per foot of shaft. Guides 4 by 6 inches in size are made from local red fir or spruce and are fastened to the end plates and dividers with 5/8 by 8-inch lag screws. Bearing sets are installed at intervals of 72 feet. The hoisting bucket and crosshead for sinking are operated in the manway

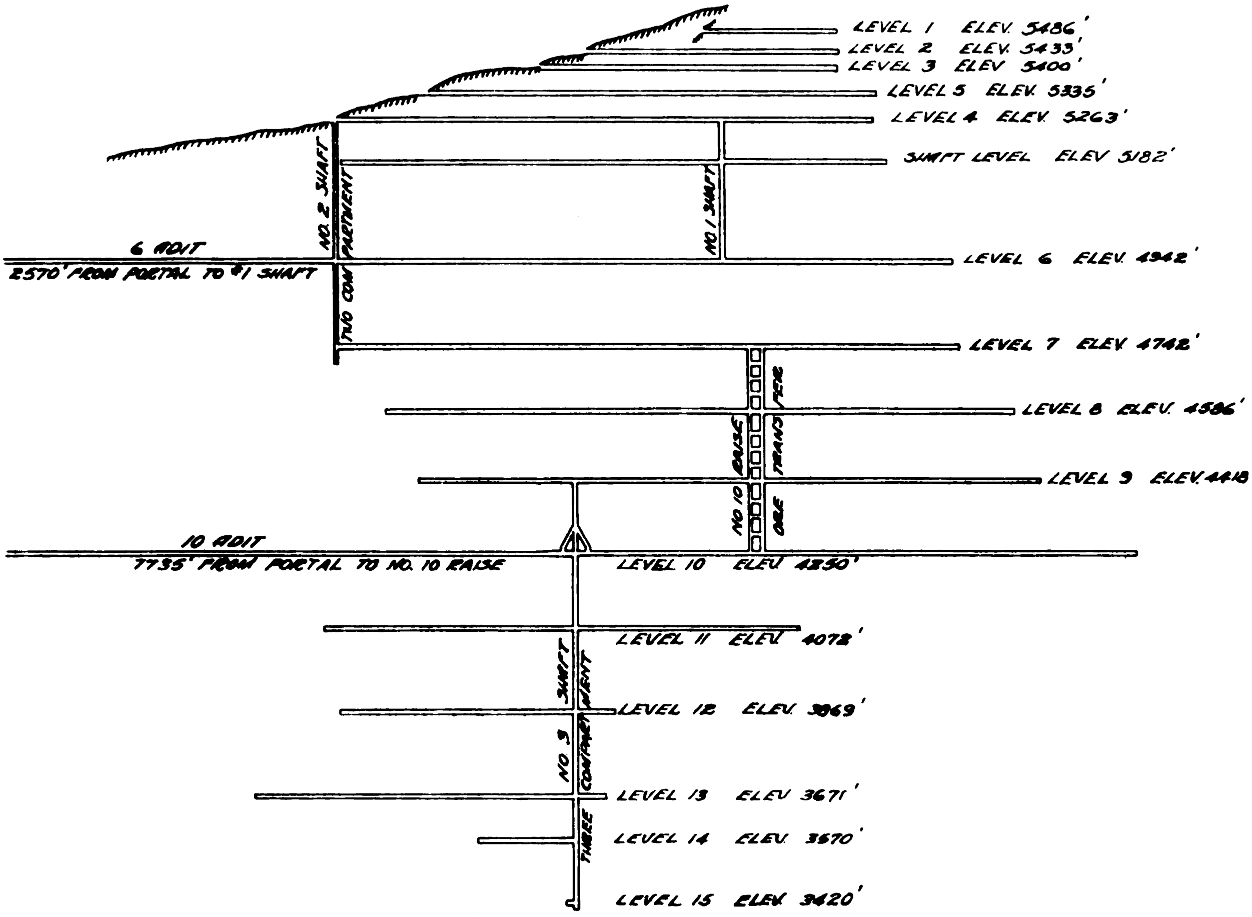


FIGURE 1.-DEVELOPMENT OF ENGEL MINE

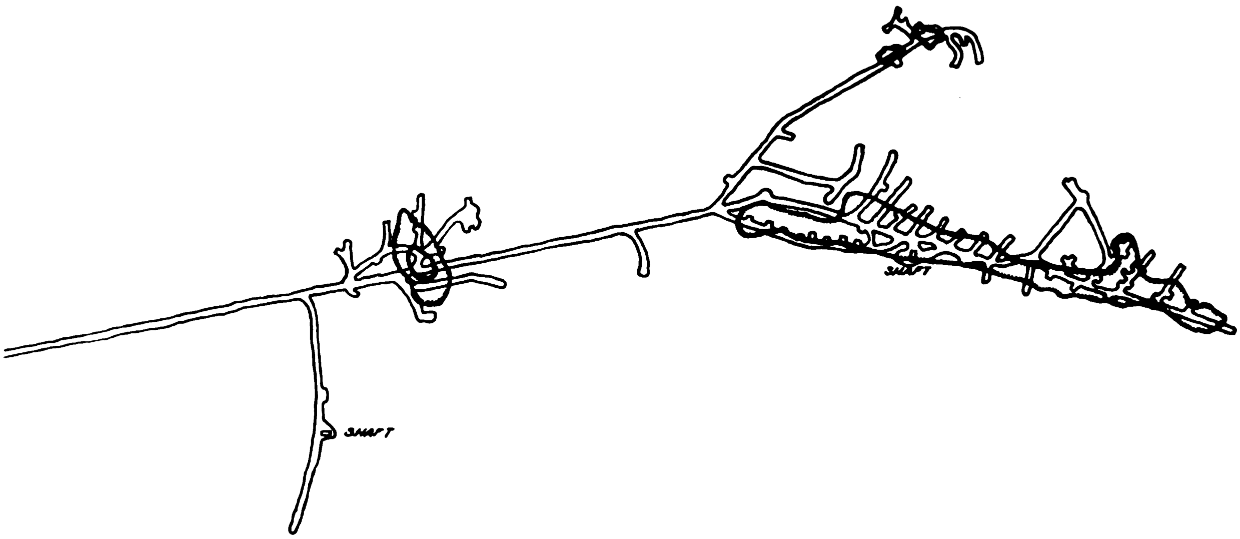


FIGURE 2.-PLAN OF LEVEL 6

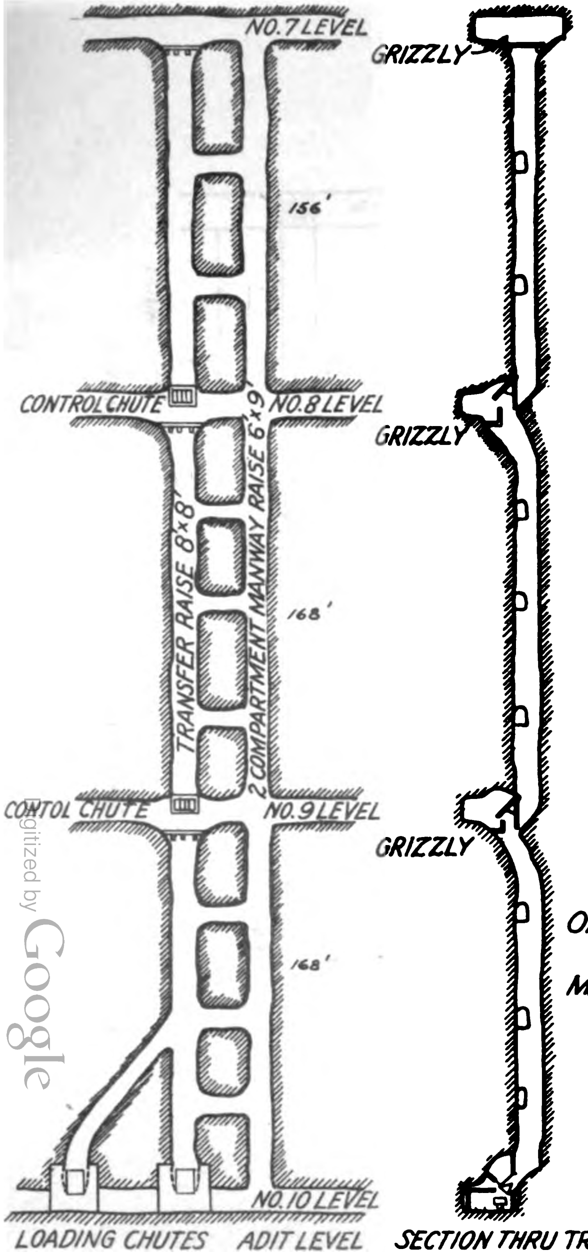


FIGURE 3.
ORE TRANSFER
AND
MANWAY RAISE

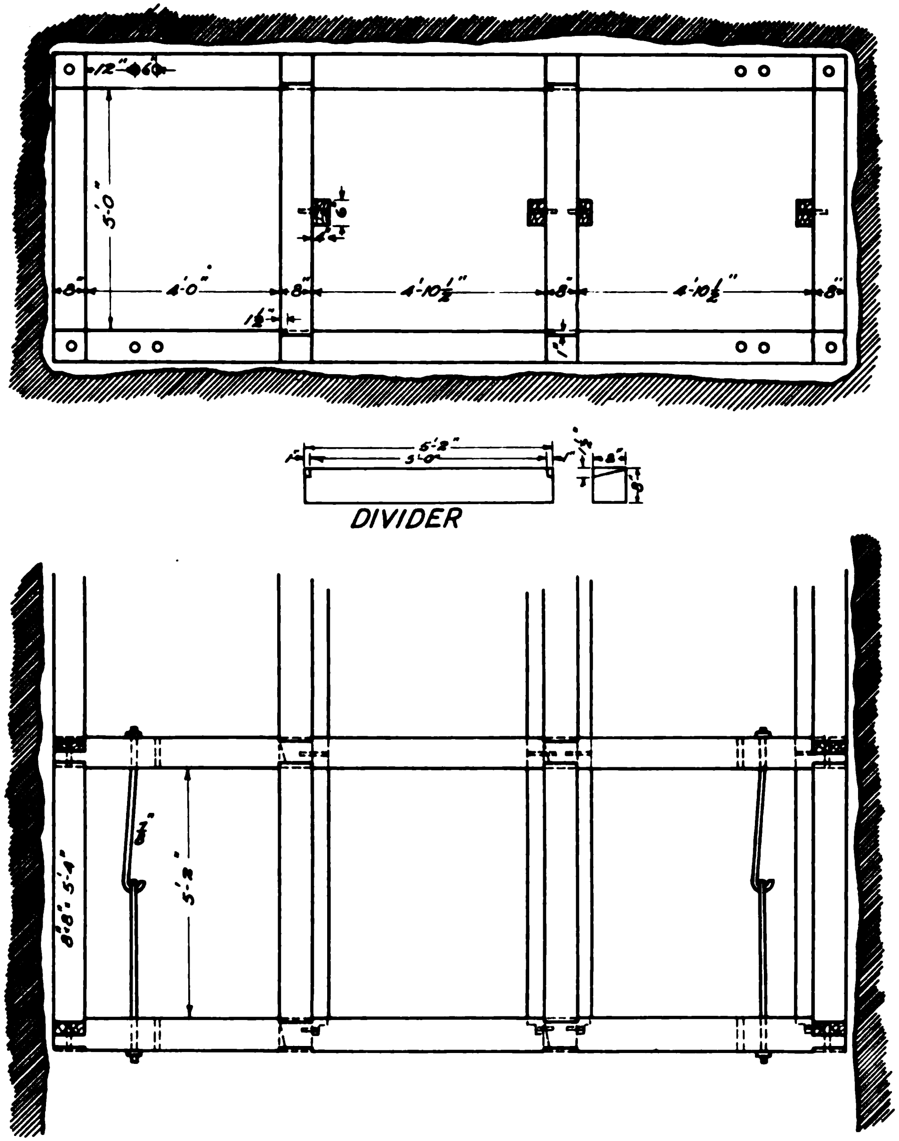


FIGURE 4.-SHAFT TIMBERING

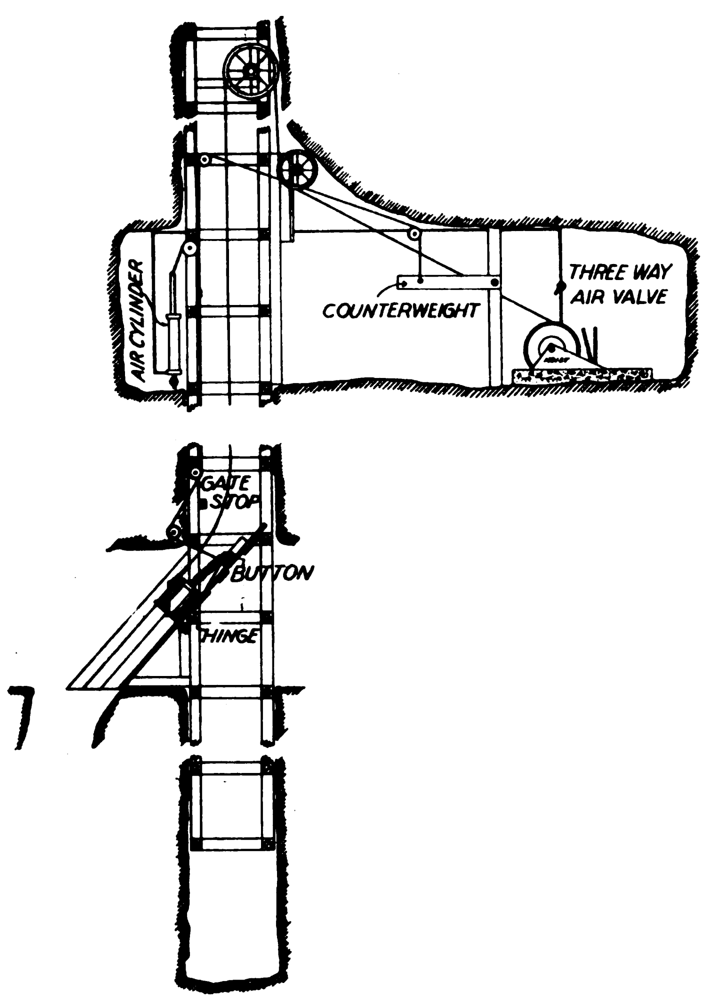
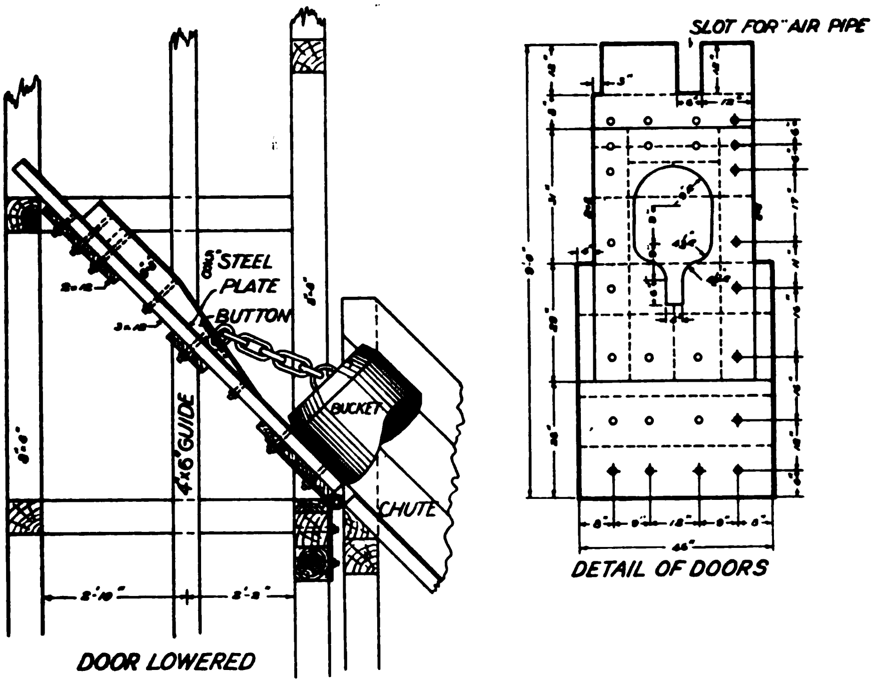


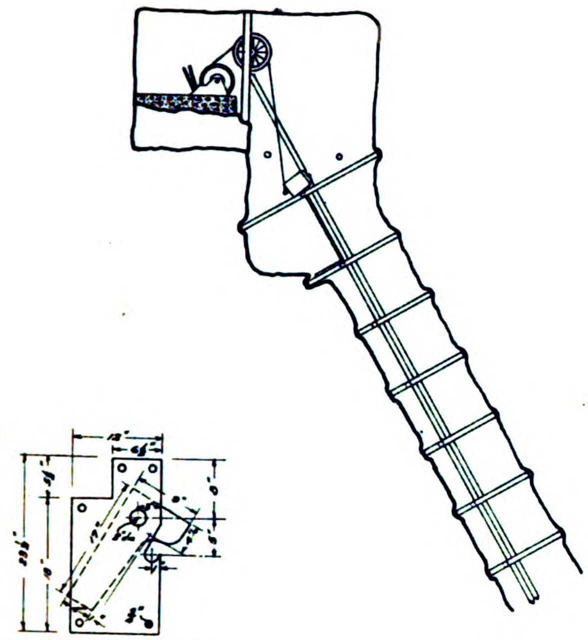
FIGURE 5.- GATE USED IN SINKING NO. 3 SHAFT



DOOR LOWERED

DETAIL OF DOORS

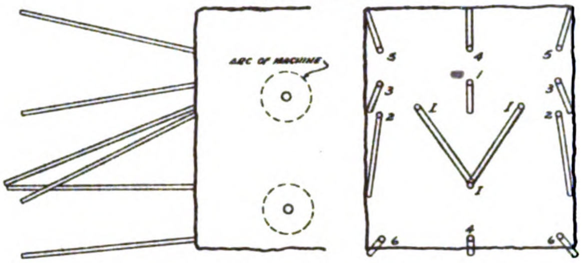
FIGURE 6.- DOOR USED IN SINKING NO. 3 SHAFT



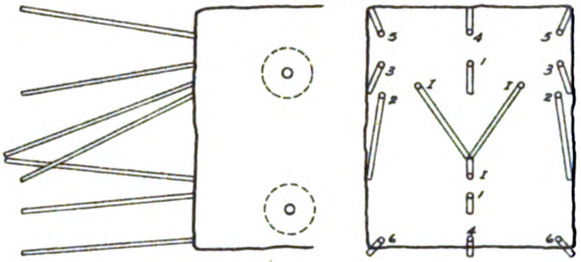
DETAIL OF BUCKET TRIP

FIGURE 7.-BUCKET ARRANGEMENT FOR INCLINE SHAFT

A-14 HOLE ROUND



B-15 HOLE ROUND



C-18 HOLE ROUND

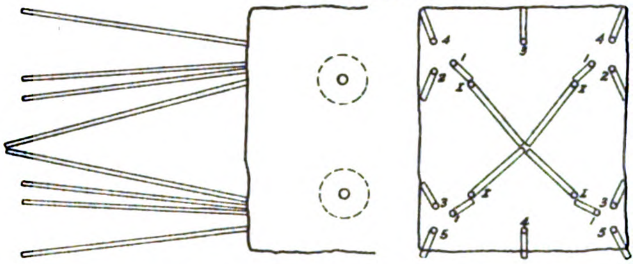


FIGURE 8.-ROUNDS USED IN DRIFTING

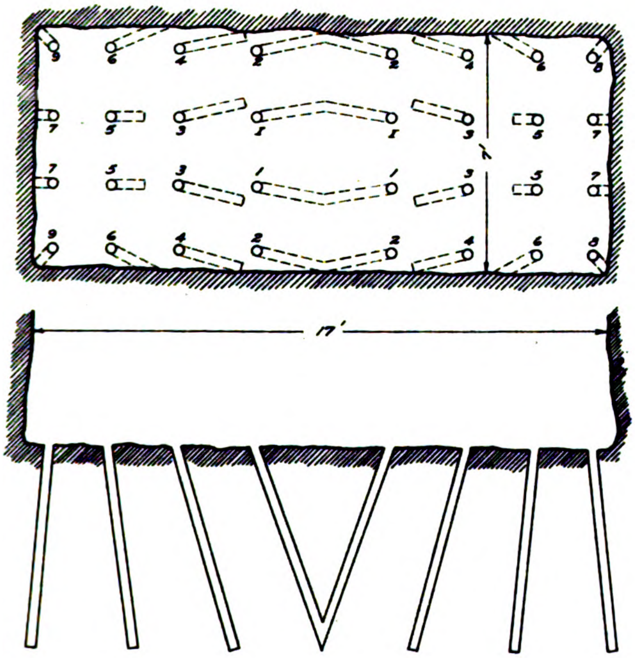


FIGURE 9.-SHAFT ROUND

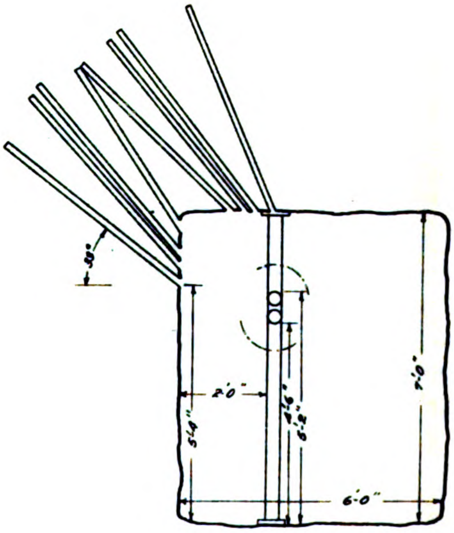


FIGURE 10
FIRST ROUND FOR CUTTING POCKETS, 18 HOLES

compartment. At present (Sept., 1929) the sinking hoist at No. 3 shaft is installed on the twelfth level and sinking is in progress 435 feet below. The bucket is dumped at the fourteenth level into a pocket from which the broken rock is drawn into the 2-ton skips and hoisted to the tenth level. The sinking bucket is 28 inches in diameter by 28 inches high and is made of 3/16-inch steel. A tail chain is 3 feet long and made of 7/8-inch iron; a 3/8-inch plate 8 inches in diameter is secured to the free end of the chain. The other end of the chain is fastened to the bottom of the bucket. When the bucket is hoisted for dumping, a door manipulated by the hoistman by means of an air cylinder and cable is dropped across the shaft, as shown in Figure 5. The door is made of 3 by 12 inch plank covered by a 1/4-inch steel plate raised 8 inches above the plank at the upper end, as shown in Figure 6. The plate curves to join the plank at the lower end and has a slot in the raised portion to engage the button or plate, causing the bucket to dump when it is lowered. If the chain should break, the plank would keep any loose parts from falling down the shaft and injuring the workmen.

Shafts are sunk on contract. The crew consists of four men each on three shifts. The contractor usually works on day shift. Fifty-five pound jackhammers, each equipped with a line oiler, are used in drilling. The drill steel is 1-inch hollow-hexagonal with 4-1/4-inch shank. The bits are the regular cross bits with the standard double taper. The starters have a gage of 2-1/2 inches to permit the insertion of a short piece of 2-inch pipe in the collar of the hole to keep out dirt and water for the remainder of the drilling. The next steel has a gage of 1-7/8 inches, and for each change thereafter the gage is reduced 1/16-inch. The starters are 24 inches long and succeeding pieces increase 15 inches for each change. Cut holes are put in about 7-1/2 feet deep and the remaining holes about 6 feet deep. Usually 28 sets of steel, five lengths to a set, are required per round. The shaft round consists of 32 holes as shown in Figure 9. The time required to drill a round is as follows:

1 hour to get equipment ready for drilling,
5 hours drilling,
3/4 of an hour to remove machines and tools,
3/4 of an hour to load and blast.

In blasting, two boxes of 60 per cent strength dynamite are required per round, or an average of about 7 sticks per hole. Electric delay detonators are used and are inserted in the third stick from the bottom of each hole. The blasting circuit consists of No. 18 insulated wire leading down the shaft from a blasting switch connected to a 220-volt circuit at the level. The cut holes are blasted in pairs to prevent the breakage of timbers. The primers are connected in parallel to two bare wires stretched across the shaft. The primers required per round are as follows (fig. 9):

2 Instantaneous
2 No. 1 Delay
4 No. 2 Delay
4 No. 3 Delay
4 No. 4 Delay
4 No. 5 Delay

4 No. 6 Delay
4 No. 7 Delay
2 No. 8 Delay
2 No. 9 Delay

The two last holes are fired on the side away from the bucket to throw the broken rock closer to it and save handling. Two shifts are required to remove the broken rock after a blast. The drilling and mucking cycle is repeated until enough progress is made to place two sets and still have about 8 feet between the shift timbers and the top of the broken rock. One shift is required to place the two sets and move the bulkhead preparatory to drilling. The bulkhead consists of 8 by 8 inch timbers covering two compartments of the shaft; it protects the sinking pump which is directly above it from flying rock and also provides storage space for machines and equipment. In timbering operations the blasting set is lowered by means of the hoist cable on one end and chain blocks on the other. The blasting set is held in place by 3/8-inch sling chains with grabhook ends. The wall plates are then lowered and suspended by 3/4-inch hanging bolts. The end plates, dividers, and posts are put in next, the hanger bolts are drawn up and the timbers wedged in place. The operation is repeated for the next set. The bulkhead is then moved down to the bottom set, and the guides are installed for the crosshead and bucket. The sinking pump is lowered to the bulkhead; pipes are extended when there is sufficient room for a length. The working face is lighted by four 100-watt lamps attached to a rubber drop-cord suspended from a reel.

In sinking inclined shafts or winzes, a sinking bucket is used that has a 2-inch rod projecting 6 inches beyond the sides fastened to the bottom. This rod is engaged in a notch at the dump and allows the bucket to tip over when the cable is lowered. The bucket is then raised over a trigger which causes the rod to slide over the notch when lowered. The arrangement is shown in Figure 7.

Drifts and Crosscuts

Drifts and crosscuts are driven 6 by 7 feet in section with an 8 by 8 inch drain ditch at the side. Practically all long drifts are advanced by contract, and the miners and muckers participate equally in the earnings of the contract. When speed in driving is desired, a round is drilled with one machine and blasted on every shift. This necessitates the use of a crossbar mounting with as many holes as possible put in from the top set-up. The 14-hole round is used (fig. 8-a) when the rock is hard to drill but relatively easy to break because of its brittleness, or when drilling time is limited. Usually, however, either the 15-hole (fig. 8-b) or 16-hole round is drilled. The average footage drilled per shift in development headings in 1928 was 11.3 holes, 5.2 feet deep, or a total of 58.76 feet. If it is found upon removal of the broken rock that the top portion of the round has been drilled a little high, a reliever is drilled from the bottom set-up, half way between the bottom cut hole and the center lifter. For the top set-up the bar is placed about 2-1/2 feet below the back of the drift. The bottom set-up is made with the bar resting on a powder box on edge with two wedges on top. The bar is 3-1/2 inches outside diameter and is made of extra heavy pipe. The drifting machines have a 3 or 3-1/2 inch bore and use 1-1/4-inch hollow-round lugged steel. The machines weigh 150 pounds fully equipped. All the drill steel has the standard cross bit with the 5° and 14° tapered sides. The gage of starters, beginning with 2-1/4 inches, is reduced 1/8 inch for each change in length. The steel is made up in 18-inch changes and is the same as the steel used in the stopes, except that the shorter lengths of each change are sorted out by the tool nippers for use in the drifts.

The drifting crew consists of one miner and two muckers. It usually takes from one-half to three-quarters of an hour to bar down loose rock and set up the machine. Actual drilling takes about five hours, but there is usually some delay in removing the broken material for the bottom set-up, so that ordinarily six hours elapse between the start and the finish of drilling. Three-quarters of an hour is needed to remove the drilling equipment, load the holes, and place the shoveling sheets. These sheets are 1/4-inch rolled steel 4 by 6 feet in size. The muckers assist the miner in setting up and tearing down his machine. The broken material is shoveled into 1-ton round-bottom cars and trammed by hand to the station.

Usually in loading holes 25 sticks of 60 per cent strength extra gelatin are placed in the cut holes and the rest of the holes are loaded with a box of 40 per cent strength dynamite. Electric primers are used which are connected in series and fired from the same circuit as the stope holes. The primers are put in the third stick from the bottom. Powder is delivered to the miner by the steel nipper.

The 18-hole round is drilled only when a clean set-up can be made, and the machine is usually mounted on a cross arm attached to a vertical column. Figure 8-c shows this round as drilled from a crossbar; the difference in placing of the holes by the two methods of supporting the drill is only slight. The 18-hole round can seldom be drilled and blasted in one shift; therefore, the holes are drilled deeper - 6-1/2 to 7 feet - and the time of drilling is extended over two shifts. This procedure is followed by men drilling on company time. The 18-hole round takes about 15 pounds of 60 per cent strength dynamite for the cut holes and 60 pounds of 40 per cent for the remainder of the round. The round breaks from 5 to 6 feet, and about 1-1/2 shifts are required to shovel the broken material.

The No. 10 adit is 8 by 8-1/2 feet in cross section; it was driven under a bonus system. A distance of 7,686 feet was driven in solid rock in 699 days, an average of 5-1/2 feet per shift. The round used was drilled by three machines from a crossbar, one machine being run by the shift boss. The crew worked in two shifts; a round was drilled, blasted, and material from the preceding round shoveled on each shift. Sometimes 12 hours was required to finish the round. Four muckers worked on each shift; two men loaded a 3-ton car and pushed it past a siding where the empty cars stood, while the two others rested and waited their turn for the next car. A tool sharpener and a helper working on day shift laid pipe and track when they had finished sharpening the steel. A foreman was in charge of the work and participated in the bonus. The men were paid the following daily wage:

1	Foreman	$6.00
2	Shift bosses	5.00
4	Miners	4.50
8	Muckers	4.00
1	Tool sharpener	5.25
__	Tool sharpener helper	4.00
17		

The average progress per month was about 330 feet. The bonus started at 210 feet and was paid as follows:

From— (feet)	To—— (feet)	Amount paid per foot	Footage	Amount of bonus paid
210	219	$2.00	10	$ 20
220	229	3.00	10	30
230	239	4.00	10	40
240	249	5.00	10	50
250	259	6.00	10	60
260	269	7.00	10	70
270	279	8.00	10	80
280	289	9.00	10	90
290	299	10.00	10	100
300	309	12.00	10	120
310	319	14.00	10	140
320	329	16.00	10	160
330	339	18.00	1	18
				$978 bonus paid for average monthly advance of 331 feet.

The $978 was divided equally between the 17 men, making an average of $57.53 per month per man in addition to his regular daily wage. There were also a motorman and a dumpman on each shift who did not participate in the bonus. The adit was driven at a cost of $30.93 per foot in hard rock, not including the preliminary cost and open-cut work.

Raises

Most of the raises in the mine are run in pillars for manways to serve stoping operations. A manway raise is made 6 by 8 feet in cross section and is run at an angle of 60° or more from the horizontal. The first round for a manway raise is drilled from the drift at 45°, as shown in Figure 10. The next three rounds are also drilled with drifter machines at the 45° inclination, and after that the raise is run on a steeper angle and self-rotating stopers are employed.

In starting a raise the first three rounds and the cut holes of the fourth round are blasted before building the chute. After the chute is in place the remainder of the fourth round is fired. This procedure avoids breaking the chute timbers. The chute consists of 18-inch round timber spragged in position on 5-1/3 foot centers. The chute and manway compartments are lined with 3 by 12 inch plank nailed to the 18-inch round timbers. The plank lining is carried to within 6 feet from the face. A bulkhead of round pole lagging 8 inches in diameter is placed over the manway side of the raise before rounds are fired. The chute and manway compartments are each 3-1/2 by 6 feet in the clear. In the manway compartment a slide is constructed for transporting timber, steel, and drilling equipment to the back of the raise (fig. 11). The slide is built of 2 by 10 inch plank and is adjacent to the chute compartment. After the installation of the chute the raise is put on a contract basis.

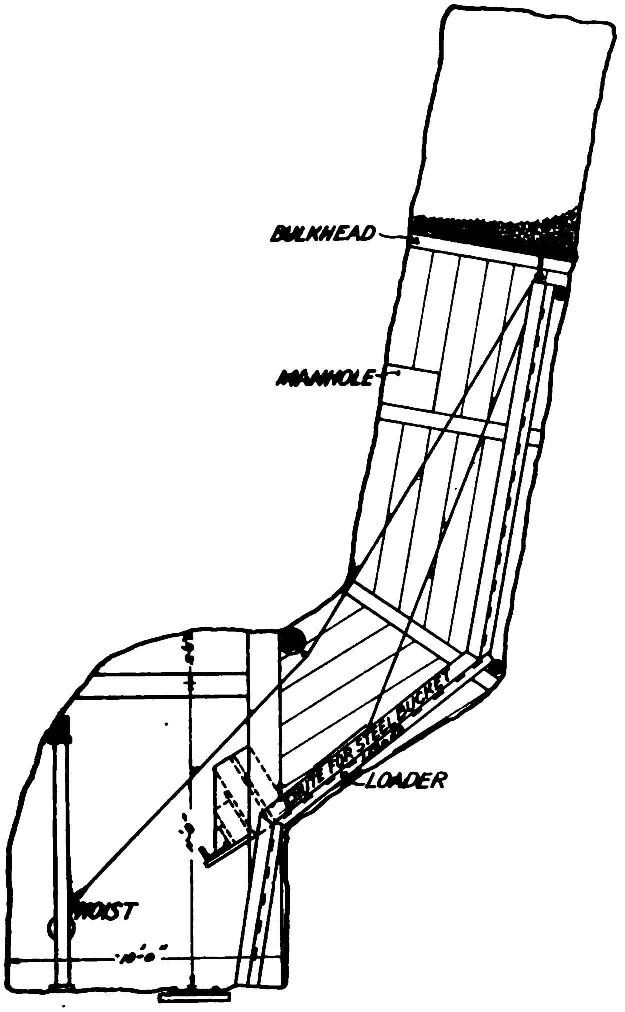


FIGURE 11.- MANWAY RAISE

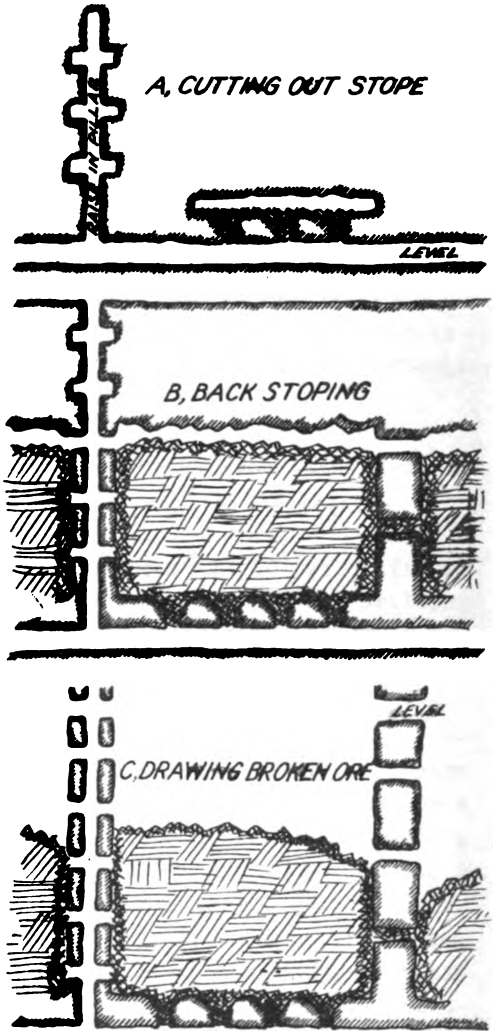


FIGURE 12.- METHOD OF STOPING

The raises are from 175 to 200 feet long. Manway crosscuts 5 by 6 feet in cross section are run on both sides of the raise at intervals of 25 feet vertically above the first crosscut. Stoping on both sides of the pillar is done concurrently with the raise work. However, owing to the danger of falling rock a raise is generally not used to serve a stope until the raise is completed. Therefore, the first crosscuts from the raise are run at the elevation that the top of the stope will have reached when the raise is completed.

Because of the gases from blasting it is customary to work only one shift per day in the long raises. Two men work on a shift; they stage-up, drill with two machines, and blast in the raise each working shift until the raise is beyond the position for a crosscut. The raise is then lined and the bulkhead built a foot below the floor elevation of the crosscut. Two drifters are set up on the bulkhead, and crosscut rounds on either side of the raise are drilled and blasted. A second round is blasted in both crosscuts before resuming the raise work. An average of 4 feet per round is made in both the raise and the crosscuts. In a month's time drilling occupies 20 shifts and timbering 10 shifts for both men. Enough broken rock is drawn from the chute each day to allow for the next blast.

Stoper raise rounds consist of from 14 to 18 holes and are blasted with electric primers. Pyramid-cut rounds similar to those used in drifting are used in the raises.

In raise work the cost of upkeep of automatic stopers is about $1.50 per shift per machine, which is considerably higher than the cost for hand-rotated machines. The drilling speed for the automatic stopers, however, is such that about an hour less is required for drilling a round. In long raises this justifies the increase in expense of upkeep over hand-rotated stopers or mounted drifters.

Figure 3 shows the arrangement of the ore pass and the manway and supply raise connecting the seventh, eighth, ninth, and tenth levels. The two raises are separated by a rock pillar 20 feet thick. At each level, with the exception of the tenth, grizzlies are installed which consist of 85-pound rail spaced so as to give 13-inch openings. At the 8 and 9 levels the ore transfer raise is offset, and a finger chute-gate is installed at each level to control the flow of ore. The finger gates hold ore in the upper parts of the ore pass when desired to provide room for dumping the ore trammed on either the eighth or ninth levels. To reopen the ore pass in the event of the ore hanging-up, drifts were driven connecting the manway raise and the ore pass raise at intervals of 40 to 60 feet vertically.

At the bottom of the ore pass two loading chutes were constructed of reinforced concrete and steel. The reinforced concrete portion of the chutes consists of a wall 1 foot thick with the necessary openings for the gate plates. Loose ground encountered where the chutes were to be built made concreting necessary. The chutes are built of sheet steel with manganese steel liners. The chute gates are of the undercut arc type operated by compressed air cylinders with four-way valve controls. To further control the flow of ore two heavy 700-pound cast-iron weights are hung in the chute on a shaft above the gate. The use of these weights enables a larger chute opening to be used, thus affording more room for barring-down large pieces of ore. The cast-iron weights were too easily broken when blasting large bowlders in the chutes, but this difficulty was overcome by jacketing the weights with sections of worn roll shell. Above the drift a wooden loading platform gives access to the chute gates.

The manway raise is 6 by 9 feet in cross section and consists of one manway and one cage compartment. The hoist for the cage is at the bottom of the raise, and the 7/8-inch hoisting cable passes up the manway and over a sheave to the cage compartment at the top of the raise.

STOPING

As previously stated, the present stoping practice in the main ore body is to leave pillars 20 feet thick across the width of the stope at intervals of 90 to 120 feet. Manways with crosscuts to the stope every 25 feet vertically are driven in alternate pillars (fig. 12). The pillars are located vertically over the pillars of the stope below. Stopes outside of the main ore body are mined without pillars.

Ore is drawn from the stopes through chutes placed 30 feet apart in a drift along the footwall of the ore body.

The three stages of work in shrinkage stoping, i.e., cutting out, back stoping, and drawing the finished stope, must be kept going continuously to maintain a uniform output from the mine. Development work must be kept far enough in advance not to hold back work in the stopes. When drifts and crosscuts are advanced far enough on a level the raises for the manways and chutes are started. At the chutes the drift is cut out to a height of 14 feet above the rail. The chute raises are started in the same manner as the starting of the manway raises previously described. Standard chutes (fig. 13) are installed. After the chutes are in place the chute raises are extended to a height of 32 feet above the rail. The raises are then belled out and connected with each other as shown in Figure 12-c. An 18-foot pillar is left between the top of the drift and the bottom of the stope. Practically all the ore is withdrawn in cutting out, leaving a room the width of the ore body and 7 feet high. One of the chute raises is used as manway through which air and water lines and supplies are taken into the stope until the back of the stope reaches a crosscut of a regular manway. After cutting out operations are completed, stoping in the back is started. Rounds are drilled with drifters as shown in Figure 14. In flat backs two rounds are drilled in opposite directions from the same set-up. These rounds consist of a single row of three holes in narrow stopes drilled at an upward angle and four holes in wide stopes. By this procedure two rounds can be drilled and blasted without burying any oversize bowlders. Breast rounds consist of two rows of holes drilled as shown in Figure 14-b. The average depth of the holes in the stopes is about 7.7 feet. During the year 1928 an average of 7.3 holes totaling 56.2 feet was drilled per man-shift in stoping. The stope rounds are blasted with 40 per cent strength ammonia dynamite in 1-1/8 by 8 inch cartridges. Seven cartridges is the average number used per hole.

Stemming in cartridge form is supplied for blasting all rounds. It consists of a mixture of sawdust and clay. The mixture is poured into the paper shells dry and then dampened with water. The sawdust holds the moisture until it is absorbed by the clay. The stemming shells are red to distinguish the stemming from the powder.

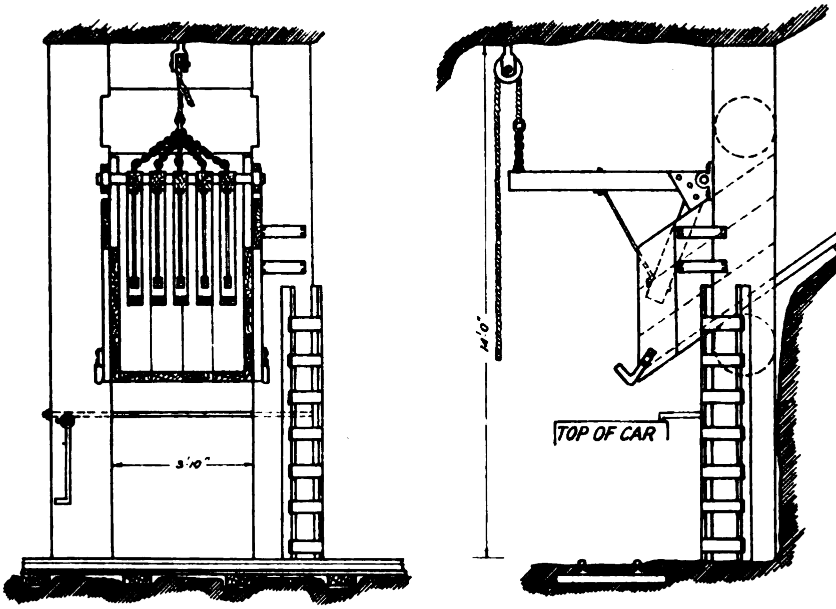


FIGURE 13.-STANDARD CHUTE

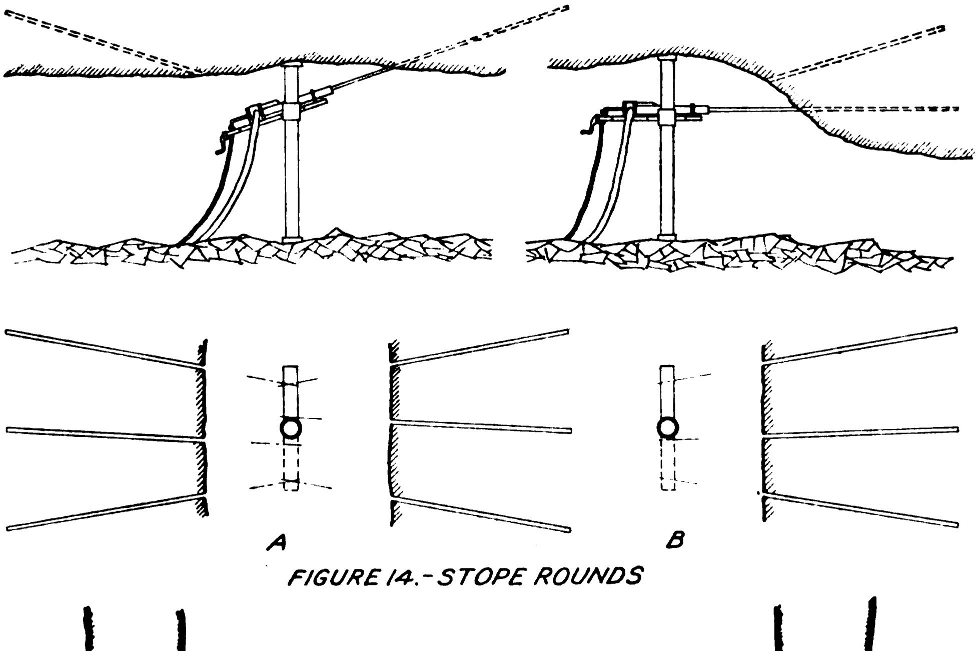


FIGURE 14.- STOPE ROUNDS

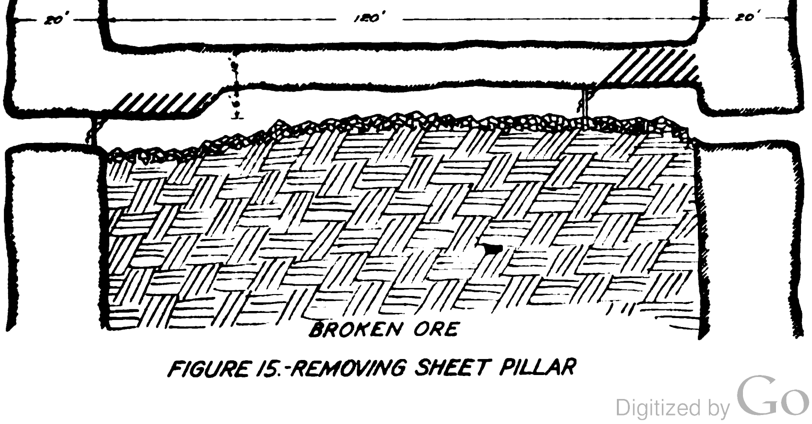


FIGURE 15.-REMOVING SHEET PILLAR

The labor and material cost of excavating a chute raise and station is as follows:

	Labor	Material	Total
Miner, 10 shifts	$50.00	-	$50.00
Mucker, 10 shifts	45.00	-	45.00
Explosives	-	$36.63	36.63
Totals	$95.00	$36.63	$131.63

All chutes for the stopes are equipped with finger-type gates, and each chute has an opening 3 feet wide and 2 feet high when the fingers are fully raised (fig. 13). The labor and material required and the cost of installing a stope chute with finger gate are as follows:

	Labor	Material	Total
Iron lining and brace	$3.44	$6.75	$10.19
Lumber	4.29	3.75	8.04
Chain	1.37	.96	2.33
Eyebolt	.52	.30	.82
Sheave	1.72	scrap	1.72
Shaft 2-11/16 inches by 6-1/2 feet	-	8.78	8.78
Shaft housing	2.03	2.65	4.68
Shaft housing bolts	-	1.00	1.00
Two shaft collars and 4 washers	1.37	scrap	1.37
40 feet round timber at 8-3/4 cents per foot	-	3.50	3.50
60 feet round timber at 6 cents per foot	-	3.60	3.60
384 bd.ft. 4 by 12 inch timber at $25 per M	-	9.60	9.60
3/16-inch steel for bottom	-	6.40	6.40
3/16-inch sheet steel for cap and sides	-	6.91	6.91
Rod 3/4-inch by 8 feet	.34	.60	.94
Cap hooks	.69	scrap	.69
Stop board	.69	-	.69
15 feet 3/4-inch rope	-	.53	.53
Labor building chute, timberman and helper	70.00	-	70.00
Caps and powder for hitches	-	2.00	2.00
Totals	$86.46	$57.33	$143.79

The crew in a stope consists of a miner and a mucker for each face in cutting-out operations. After back stoping is started one to three miners and one or two pluggers work in each stope, depending on the size of the stope. The pluggers help the drillers to set up, and spend the rest of the time in plugging large bowlders and in breaking the smaller oversize fragments with a hammer. No blasting chambers are used over the chutes, and as many of the large pieces as possible are broken in the stope after being blasted down.

On day shift the drillers bar down the back, set up their machines and drill during the rest of the shift. No holes are blasted on day shift, except when, as occasionally happens, the round has misfired on the preceding shift. The miners on the night shift continue the drilling, tear down the machines, remove the equipment to a safe place, and load and blast the holes.

A 2-inch pipe is used for air and a 1-inch pipe for water. These pipe lines have short lengths of rubber hose inserted at every other joint to give flexibility to the pipe lines. A pipeman and a helper lay the pipe in the stope at the beginning of each day shift.

The average routine machinemen in the stopes is as follows:

Operation	Day shift, hours	Night shift, hours
Barring-down........	2	-
Setting-up...........	1	-
Drilling................	5	6
Tearing down and removing equip.	-	1-1/4
Loading holes.....	-	3/4

Electric Blasting

Stope rounds are blasted electrically; the primers are connected in series. The plugger holes are also connected in series and both circuits are connected in parallel to lead wires extending from the stope but not connected to the blasting line. When loading is completed the lead wires are connected, and the shift boss, after checking over the men to see that all are at the station at the end of the shift, throws in the switch which sets off all blasts on the level. When electric blasting was first started a blasting crew loaded and blasted all holes between the night and the day shifts. This practice has been changed to that of having the miners load their own holes. The blasting circuit is 220 volts. A switch box shown in Figure 16 is located at the station of each level. The box is made of 18-gage galvanized iron and contains a 2-pole knife switch with fuse plugs in series and with a handle so constructed that the door of the box can not be closed and locked unless the switch is open. Only the shift boss carries the key, and as an added safety precaution he usually carries the fuse plugs in his pocket also.

Removal of Sheet Pillars

The stopes are carried to within 20 feet of the level above, at which point backstoping is discontinued until the stope above has been emptied. The back is then mined up to the level in the usual manner. At the level the angle of the holes is changed from nearly flat to 45°. The first cut, as shown in Figure 15, is drilled with holes 9 feet deep and 4 feet apart. The holes are blasted in groups of 10 or 20 at a time, according to the ground; the bench is carried the length of the stope. The next cut brings down the remainder of the pillars. In weak ground the second cut follows closely behind the first one.

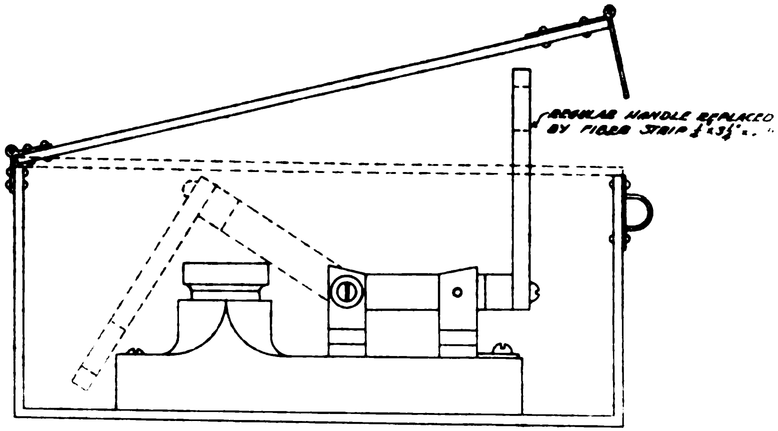


FIGURE 16.-ELECTRIC BLASTING SWITCH
D.P.S.T. SWITCH AND FUSE BOX

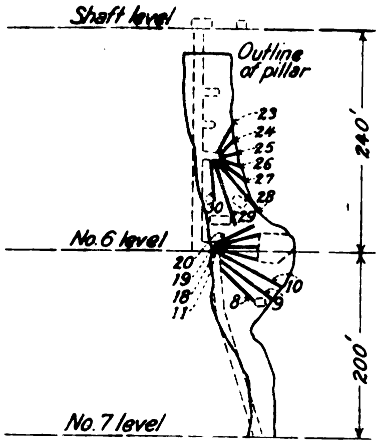


FIGURE 17
POSITION OF DIAMOND DRILL HOLES IN PILLAR

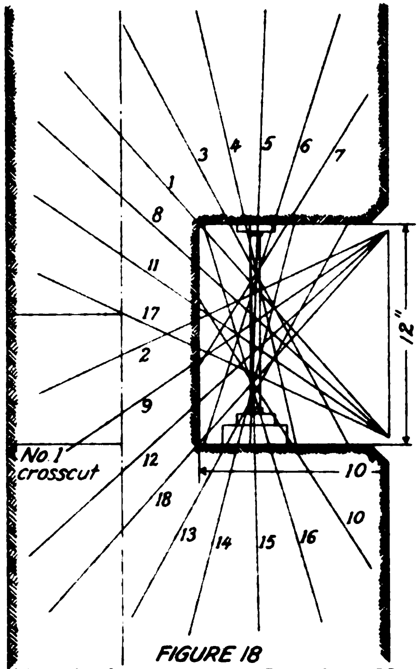


FIGURE 18
METHOD OF DRILLING VERTICAL PILLARS

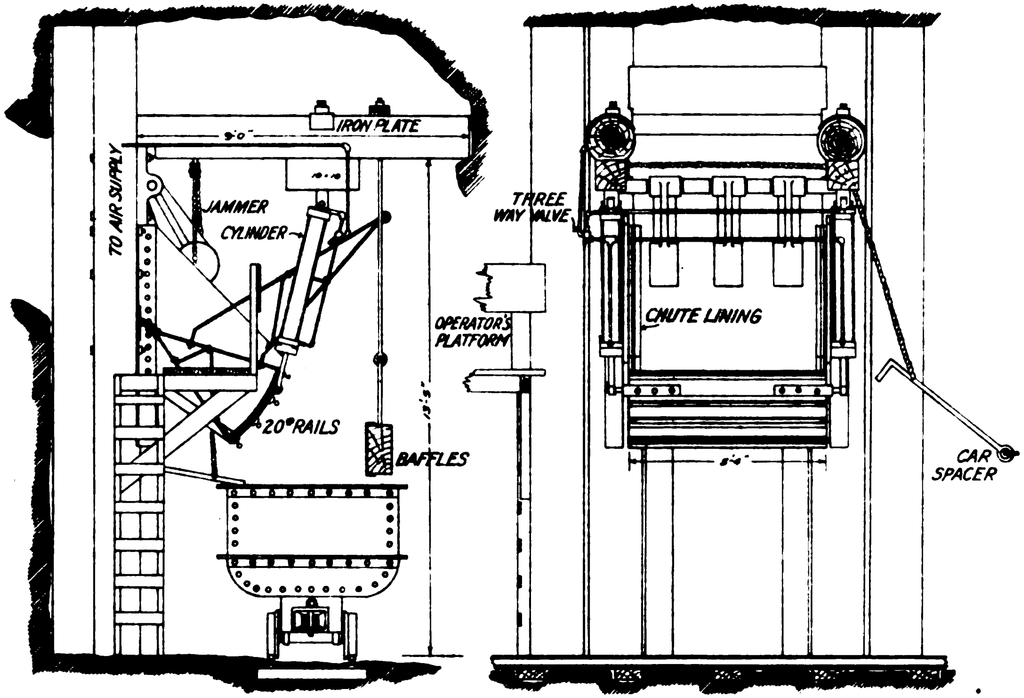


FIGURE 19.-ARC GATE CHUTE

In removing a sheet pillar from the level to the bottom of the stope above, an average of 75 tons is broken per machine shift as compared with an average of 27 tons for shrinkage stoping.

No attempt is made to break up the oversize in the stope after the last bench is shot. This results in a lot of large bowlders that must be shot in the chutes when the broken rock is nearly all drawn from the stope. As the bowlders hang-up some distance above the mouth of the chute, it is necessary to use powder on the smaller fragments around the edges of the bowlders in order that they may be brought down and drilled in safety. Powder consumption is therefore considerably higher in chute blasting during the last stage of drawing than at first.

Removal of Vertical Pillars

Vertical pillars containing a total of 146,526 tons have been removed between the tenth and the shaft levels at a cost of 58 cents per ton for ore broken by air-machine drilling and of 28 cents per ton for diamond drilling. Of the above tonnage, 19 per cent was broken by machine-drilled holes and 81 per cent by diamond-drilled holes.

The removal of vertical pillars was made possible by the firmness of the walls of the emptied stopes between the upper sheet pillar at the shaft level 300 feet from the surface, and the lower sheet pillar at the 10 level 932 feet below. As a precautionary measure against dilution in the event of caving, all of the stoping ore is extracted before the removal of the vertical pillars. The pillars are then removed one at a time and most of the resulting ore is drawn from a stope before the next pillar is blasted.

In the removal of the vertical pillars machine drilling was first tried. Some of the pillars had 6 by 8 raises with manway crosscuts at intervals of about 35 feet vertically. These pillars were attacked first; benches were cut at each manway crosscut across the width of the pillar, beginning at the upper manway crosscut. The benches were all cut on the same side of the pillar, as inclined cleavage planes caused the rock to break to smooth sloping surfaces, making it impossible to maintain a safe working floor on the opposite side of the pillar.

In starting a bench the first holes drilled from the manway crosscut were parallel to the face of the pillar, and were blasted one at a time with little burden to prevent broken rock from falling down the raise. As the bench was advanced, several holes in a row were fired simultaneously. Each bench was 12 to 14 feet high and from 10 to 14 feet wide. On completion of the bench, holes were drilled from the benches with machine drills (fig. 18). The holes had an average depth of 11.5 feet and were spaced so that the bottoms of two adjacent holes were about 4 feet apart.

The drill steel had an average run of 18 inches. Starters were 3-1/2 feet long with 2-1/4 inch gage. The gage on succeeding bits was reduced 1/8-inch for each change of steel up to the 9-1/2-foot length. From 9-1/2 feet to 14 feet the gage was reduced 1/16-inch per change, making the bit on the 14-foot steel 1-9/16 inches.

The up-holes were drilled without difficulty, but trouble occurred in drilling the down-holes as the long steel stuck when broken ground allowed the water to escape. At the start, one machine operated by a miner and a helper worked on a bench; later two machines per shift were run on the same bench by two miners who assisted each other in setting up the machines and in changing the long steel.

In blasting the upper part of the first pillar, 828 holes were fired simultaneously. The holes were loaded with 5,500 pounds of 60 per cent strength gelatin dynamite and were detonated with instantaneous electric detonators. Groups of 10, 20, 30, 40, and 50 primers in series were connected in parallel to the main lead wires. The lower numbered series in each case were placed in the outside holes of the pillar to take advantage of any time interval that might exist between the series. This method was used in preference to that employing parallel groups of 50 primers in series, to avoid the possibility of an inside group of holes firing before the outside. A 440-volt circuit was used to ignite the primers. The pillar broke clean, and only a few large bowlders were observed in the stope after blasting.

After the first pillar was removed, a second pillar 440 feet high was attacked at the end of a stope. In this pillar the manway crosscuts were on only one side of the raise for a distance of 200 feet. At a point 100 feet from the top of the pillar a blast in cutting out a bench knocked off a large slab on the opposite side of the pillar. This left the bench in an unsafe condition for further working. Furthermore, a little lower than this bench, stoping operations had removed a portion of the pillar on the hanging-wall side, so that the continuance of benching was extremely hazardous. Diamond drilling was therefore resorted to in drilling the remainder of the pillar. With the benches already cut, sufficient room was available to set up the diamond drill and put in two sets of fanning holes. The holes were all in the same plane through the center of the pillar. Figure 17 shows the outline of the pillar and the arrangement of the holes. In pillars subsequently removed the diamond-drill holes were drilled parallel to each other 10 feet apart as shown in Figure 20. Diamond drill holes were 2-1/4 inches in diameter and were drilled on contract at $3.50 per foot. In some cases suitable stations could not be obtained for parallel holes, and the fan holes 10 feet apart at the bottom had to be used. In drilling inclined holes the machine was adjusted to the desired angle for spacing the holes by means of a clinometer held on the drill rod. The drill holes were platted on a vertical transverse section of the pillar drawn on paper.

When necessary to drill through a crosscut by the diamond drill, casing was inserted at the crosscut. The lengths of the holes ranged from 20 to 207 feet for down holes and 20 to 183 feet for up-holes.

When drilling in the previously mentioned pillar was completed, there were 1,835 feet of machine-drilled holes averaging 11.9 feet deep and 741 feet of diamond drill holes averaging 49.4 feet deep. The machine-drilled holes were loaded with 60 per cent and the diamond-drilled holes with 90 per cent strength gelatin dynamite. Electric primers and cordeau were used to detonate the respective holes. Preparing benches on this pillar broke 4,472 tons and the final blast broke 20,882 tons.

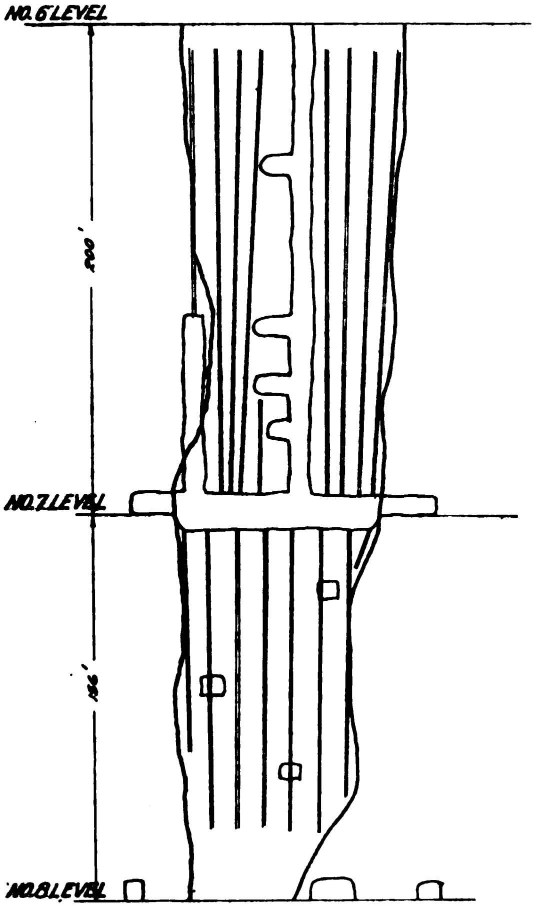


FIGURE 20.- PARALLED DIAMOND DRILL HOLES FOR REMOVING VERTICAL FILLER.

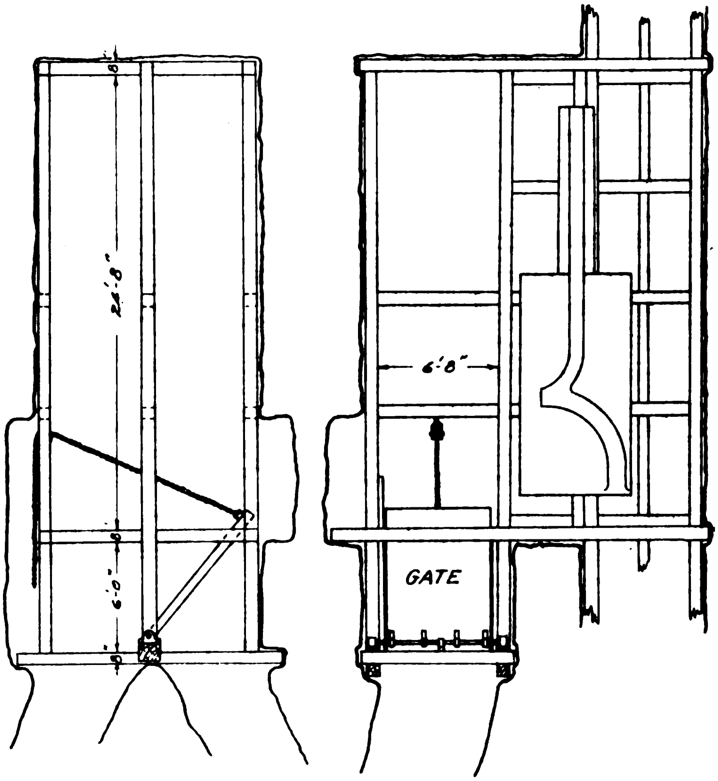


FIGURE 21.- SKIP DUMP STATION

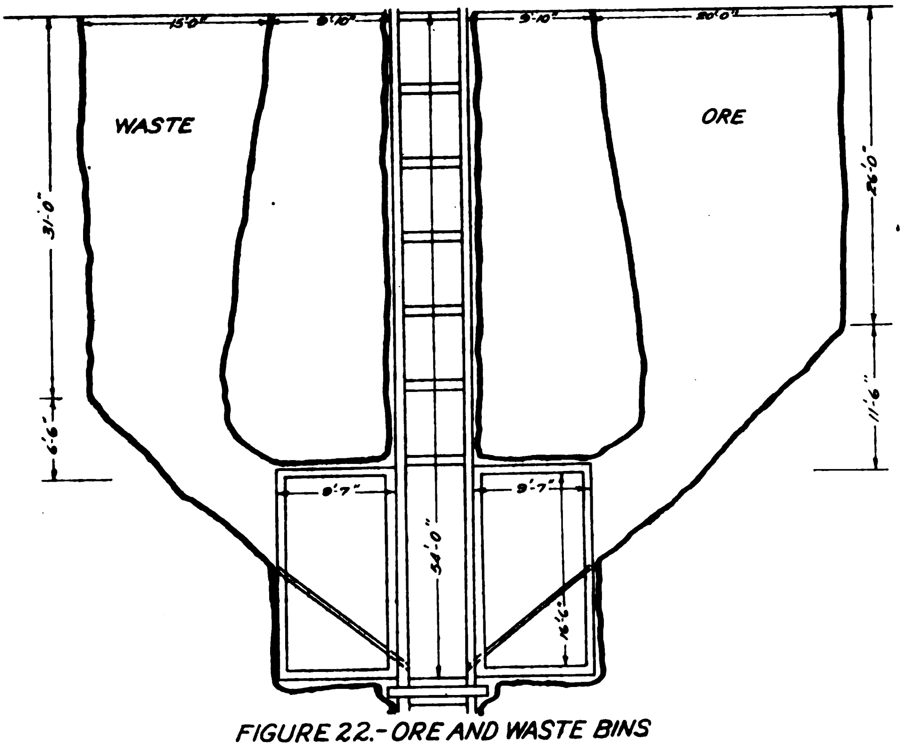


FIGURE 22.- ORE AND WASTE BINS

On account of the great length of some of the diamond-drill holes, special equipment was made for loading and tamping the explosives. For down holes a 10-foot length of brass pipe weighted inside with an iron rod was used as a tamping tool. A section of shovel handle was screwed into the bottom end of the brass pipe. The wood portion of the rod was protected from wear by a piece of rubber air hose stretched over the end. In loading, two cartridges at a time were dropped into the hole and were followed by the tamping rod suspended by a 1/4-inch flexible cable wound on a reel. One man raised and lowered the rod by windlass and the other man dropped the cartridges into the hole and jogged the tamping rod several times after it had reached the bottom. The number of turns on the windlass indicated whether or not the hole was being loaded properly. When properly tamped the 2-1/4-inch diameter holes held about four cartridges, each 1-1/4 inches in diameter and 8 inches long, per foot of hole.

Tamping rods for up-holes were made of red fir or spruce cut in 10-foot lengths. The ends of the rods were fitted with brass joints. A 1/2-inch tapered knob projecting inward on the outside sleeve kept the joints from coming apart and in addition enabled the joints to be quickly connected or disconnected. The upper end of the first rod to enter the hole was fitted with a threaded piece of 1-inch brass pipe. This pipe held either a piece of hardwood for tamping or any one of the various fishing tools that were used. The speed of loading holes was increased by means of a device to hold the rods while attaching or detaching a length. It consisted of a tripod with legs of 1-inch pipe; the tamping rods pass through a 2-1/2-inch pipe fastened to a plate on the tripod and are gripped by a cam actuated by a counterweighted lever. The rods are securely held while a section of rod is being removed or attached. The rods are released by raising a lever attached to a rope which passes over a pulley to a foot pedal. The long up-holes require a crew of three men to handle the rods.

In loading up-holes four cartridges were joined together by wooden pegs inserted in the ends. The principal trouble in loading both up and down holes was caused by broken rock working into the holes. In ravelly down-holes thin slip-jointed tubing was used. The powder and tools were lowered through the tubing and the tubes were withdrawn by cable attached to the bottom joint as loading progressed. The time and the amount of dynamite required for loading up and down holes where little delays were encountered is shown in the following tabulation:

Time and amount of dynamite required for loading up and down holes where delays were encountered

Depth of down holes feet	Number of cartridges loaded into hole	Number of cartridges per foot of hole	Time required to load hole, hours
72	290	4.0	4
83	344	4.1	4-3/4
118	493	4.2	7
Depth of up holes, feet			
180	811	4.5	30
173	726	4.2	24
181	747	4.1	32

The amount of chute-blasting required for run-of-mine ore from back-stoping in comparison with the blasting necessary in the same chute for ore broken by diamond-drill holes in the vertical pillars is shown as follows:

	Back stoping	Vertical pillars
Cost per ton for explosives and primers..	$.0154	$.0832
Number of tons drawn from chute per shot	14.91	5.74
Number of cartridges per shot....................	2.8 (mostly 25%)	6.2 (mostly 40%)

As shown in the foregoing table, the ore from the vertical pillars broke much larger, necessitating a greater amount of blasting to enable it to be drawn through the chute. However, the oversize bowlders from back stoping were plugged in the stopes and any that were missed were plugged later in the chutes. No bowlders from the pillar ore were drilled in the stope or at the chute because of the possibility of missed holes. No missed holes, however, have been discovered to date.

The diamond-drill method was found to have the following advantages over machine drilling in extracting vertical pillars: The factor of safety was much higher, since the danger from falling rock could be practically eliminated as only a single bench is required for the diamond drill. In machine drilling each bench cut weakened the pillar and increased the tendency of the pillar to slough between benches. Once a bench was cut for diamond drilling there was no need for the men to work near the edge of the pillar where there is danger of slipping or falling over the edge into the stope. Cost per ton of rock broken was much lower, and the time required to drill and blast was considerably less. It is believed that the amount of blasting required at the chutes is less for the pillars broken by the diamond-drill holes than by machine-drilled holes from benches. The following data show the comparative cost of breaking ore in vertical pillars by the two methods:

Comparative cost of breaking ore in vertical pillars

Method	Tons in pillar	Number of holes	Length of holes	Total[1] cost of drilling	Total cost of loading and explosives	Total cost per ton of broken ore
Machine drill	7,475	1,205	12,144	$2,911.00	$1,439.40	$0.5819
Diamond drill	40,178	17	2,208	8,815.00	2,052.14	.2704
Diamond drill	11,205	6	688	2,446.00	289.62	.2441

[1] - Total cost of drilling includes estimated cost of sharpening and replacing worn steel and the upkeep of machine drills, and cost of labor for drilling.

UNDERGROUND TRANSPORTATION

Tracks on the main level consist of 20-pound rails laid on 4 by 8 inch ties 3 feet long. The gage of the track is 18 inches, and the grade is one-fourth of 1 per cent in favor of the loaded trains.

Two 3-ton storage-battery locomotives are used for hauling trains on the haulage levels. A train consists of 10 one-ton round-bottom cars on all levels except the tenth. Some hand-tramming is also done from the stopes. The 1-ton cars require two men to dump. This is done by the brakeman and a man who works on the grizzly at the dumping pocket. The loading crew consists of three men who, with the assistance of the motorman, can load about 100 cars in a shift. The leader of each loading crew is a chute blaster who is responsible for the loading. In loading the cars, the smaller of the fragments too large to pass through the chute gate are bulldozed before they come up against the fingers of the gate. The large oversize bowlders are drilled with a stoper using long steel through the fingers of the gate. The holes thus drilled are blasted with 40 per cent strength ammonia dynamite and electric detonators.

To facilitate stopping the cars at the right point beneath the chutes for loading, a hook that drops over the top of the cars is fastened to the wall or post at each chute.

The main haulage or tenth level has a 24-inch gage track of 30-pound rails laid on 5 by 8 inch ties 4 feet long. The grade is one-fourth of 1 per cent in favor of loaded trains. On this level 6-ton cars are used. A train of 10 of these cars is pulled by a 7-ton electric locomotive. The cars are loaded from the ore pass described in the section under development or from the tenth level stopes either through direct chutes or from bulldozing chambers. Undercut arc gates are used at the ore pass and finger gates at the chutes from the stopes and bulldozing chambers. In six hours a crew of four men will load 40 of the 6-ton cars from a stope in which there is considerable oversize. Two men can usually load 10 cars from the ore pass in 10 minutes. This ore has previously passed through a grizzly.

The grizzly area at ore pockets or at the ore pass is usually 7 by 18 feet. The grizzly bars consist of 80 or 85 pound rails spaced 13 inches apart in the clear. The ends of the rails fit into notches cut in 24-inch round timbers and are held in place by bars bolted to the timbers.

All ore from below the tenth level is hoisted in 2-ton skips and dumped into bins from which loading is controlled by arc gates of the type shown in Figure 19. A large door lined with rails diverts the material from the skips into either the ore or waste loading bins (fig. 21).

Figure 22 shows the arrangement of the ore and waste bins at each level. The space at the bottom of the bins and at the side of the shaft is occupied by the gates and loading station.

With the 2-ton skips 450 tons of ore and waste are hoisted per day. As the skips are busy during the day shift in transporting timber, steel, powder, and other supplies, most of the hoisting is done on the night shift. The hoist has a rope speed of 800 feet per minute and is operated by a 150-horsepower motor. An additional 225-horsepower motor is being installed to operate two 5-ton skips in order to handle a larger tonnage. The space between grizzly rails at the skip pockets will be reduced to 11 inches to facilitate skip loading.

A typical skip chute is shown in Figure 23. The skip tender has some trouble in loading the 2-ton skips without spilling on account of the size of the rock. The air-controlled slide worked in conjunction with the fingers prevents overloading of the skip to a great extent. It is anticipated that the 5-ton skips will give less trouble from overloading.

Transportation of Men

The men are taken to and from the No. 10 raise on the tenth level in closed cars or "Pullmans" (fig. 24). Each car holds from 10 to 12 men. The forward car is equipped with a double door to keep the cold air in wintertime from blowing through the cars.

PERCENTAGE OF EXTRACTION

Above the tenth level, 2,901,180 tons of ore have been mined, and it is estimated that there are 107,728 tons of ore remaining in the pillars. This makes an extraction of 96.42 per cent. By using the same mining methods below the tenth level, the extraction will probably be at least 95 per cent from the lower levels.

WAGE SCALE AND CONTRACT RATES

The following wage scale for the principal classifications of underground employees was effective in October, 1929:

Brakemen	$4.50
Car loaders	4.50
Chute blasters	5.00
Hoistmen	5.00 and 5.75
Jackhammer men	4.75
Machinemen	5.00
Motormen	5.00
Muckers	4.50
Pipe and trackmen	5.00
Timbermen	5.00 and 5.50
Timbermen helpers	4.50
Shift bosses	6.00
Skip tenders	5.00

All employees in underground operations are paid twice a month for the time worked up to 10 days before the pay day.

Most of the development work is contracted at prices which vary according to working conditions. Six by seven drifts are contracted at $9 per foot, the contractor furnishing his own explosives and removing the broken rock to the bins at the station. Manway raises 6 by 8 feet in cross section are contracted at $9.75 per foot for the first 100 feet and are increased $1 per foot for each additional 100 feet above the level. Crosscuts from the raise are paid for at the same rate. The contractor furnishes his own explosives. The company draws rock from the raise as requested by the contractor. In sinking a 7 by 17 foot shaft the contract price is $50 per foot of shaft. The contractor furnishes the explosives required and the necessary rubber clothing. The company furnishes all other supplies and a hoistman to operate the sinking hoist.

Board at the company boarding house costs $1.25 per day, and furnished rooms in the dormitory maintained by the company cost $4 per month. Hot coffee is served underground at the various stations at lunch time.

VENTILATION

Ventilation is natural. In long drifts and in shaft sinking, blowers are used to clear out the powder gases.

SAFETY METHODS

Fire hazards are negligible, as very little timber is used in the mine. Chutes and shaft timbering comprise the main usage. The shafts are usually wet or dampened with water for the preservation of timbers.

Conditions in the mine and plans for their betterment are discussed at a general meeting of the shift bosses and foremen on the first change day of each month. A list of accidents of the preceding month is made up and the means to be taken for preventing the recurrence of similar accidents are discussed. It is desired to have the shift bosses educate the men under their supervision in safe methods of working, as well as to obtain quantity in production. The safety meetings are of considerable value in giving a general understanding of what is wanted, and they impress upon the bosses the importance of instruction in safe methods of working.

ADMINISTRATION

The superintendent sees that all departments function smoothly and in harmony, directs the mine foreman in development and mining work, and checks up on the results obtained.

The mine foreman hires the men and confers with the superintendent in regard to promotions. He directs the shifters as to location of set-up and general direction of drilling for machines; designates chutes and amount to be drawn, lays out the details of development, controls the steel-sharpening and repair work of the blacksmith, and orders all supplies.

The shifters instruct the miners where to set up; the number, direction, and depth of holes to drill; the amount of powder to use; give the trammers their instructions, and keep the time and record of performance of men underground.

The steel nippers take sharp steel to the miners and remove dull steel from the stopes. A nipper on each level issues powder at the magazine and reports amounts used and the location to the shifter. The other nippers get the order from the miners for explosives and deliver it to the miners.

The mine electrician is responsible for the upkeep of all motors and locomotives underground, as well as of the power and the phone circuits. He takes orders from the mine foreman or shifter to make minor repairs in case of breakdown and arranges with the mine foreman for time to make major repairs. Any problems that arise are taken up with the chief electrician. The mine foreman may call on the master mechanic or chief engineer for any work out of the ordinary.

TABLE 1. - SUMMARY OF COSTS

Engels Mine. Year of 1928.

Tons of ore hoisted: 303,301.

Mining method: Shrinkage with pillar support.

Underground costs per ton of ore hoisted

	Labor	Supervision	Compressed air, drills, steel, and power	Explosives	Timber	Other supplies	Total
Development	$0.266	$0.021	$0.054	$0.110	$0.015	$0.013	$0.479
Mining	.378	.033	.135	.157	.017	.040	.760
Transportation	.384	.016	–	–	–	[1].038	438
Totals	1.028	.070	.189	.267	.032	.091	1.677

1 - Includes power and timber.

Note: General underground expense and surface expense applicable to underground operations are proportioned in the above amounts.

TABLE 2. - SUMMARY OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Engels Mine. Year 1928.

Tons of ore hoisted: 303,301.

Mining method: Shrinkage with pillar support.

	Development	Mining	Total
A. Labor (man-hours per ton):			
Breaking	-	-	.642
Timbering	-	-	.099
Mucking	-	-	.266
Haulage and hoisting	-	-	.381
Supervision	-	-	.074
General	-	-	.073
Total labor	-	-	1.535
Average tons per man per shift, including surface labor chargeable to underground operations	-	-	5.21
Labor per cent of total cost	25.55	39.97	65.52
B. Power and Supplies:			
Explosives (lbs. per ton)			
25 per cent	-	.128	.128
40 per cent	.442	.593	1.035
60 per cent	.124	.055	.179
90 per cent	-	.037	.037
Total	.566	.813	1.379
Timber (ft. per ton)			
Sawed timber (bd. ft.)	-	-	1.231
Poles (linear ft.)	-	-	.074
Total power (kwh. per ton)	3.436	6.203	9.639
Supplies and power, per cent of total cost	12.28	22.20	34.48
C. Per cent of Total Cost	37.83	62.17	100.00

TABLE 3. - DETAIL OF COSTS PER FOOT FOR TOTAL DEVELOPMENT

	Drifts	Crosscuts	Raises	Shafts
Size of excavation, feet	6 by 7	6 by 7	6 by 8	7 by 17
Timbered, per cent	None	None	20	50
Property of rock	Hard	Hard	Hard	Hard
Number of feet	3,372.74	2,808.25	4,322.84	491.96
Total cost per foot	$13.09	$13.09	$18.34	$60.49

A. Labor (cost per ft.):	
Breaking	$4.82
Timbering	.83
Mucking and tramming	4.70
Hoisting	.87
Supervision	.60
Total labor	$11.82
B. Power and Supplies:	
Explosives	$3.01
Timber	.41
Breaking	1.52
Mucking and tramming	.16
Hoisting	.36
Other supplies	.22
Total power and supplies	$5.68
C. Labor, per cent of total cost	67.54
Power and supplies, per cent of total cost	32.46
Total cost per foot of development	$17.50

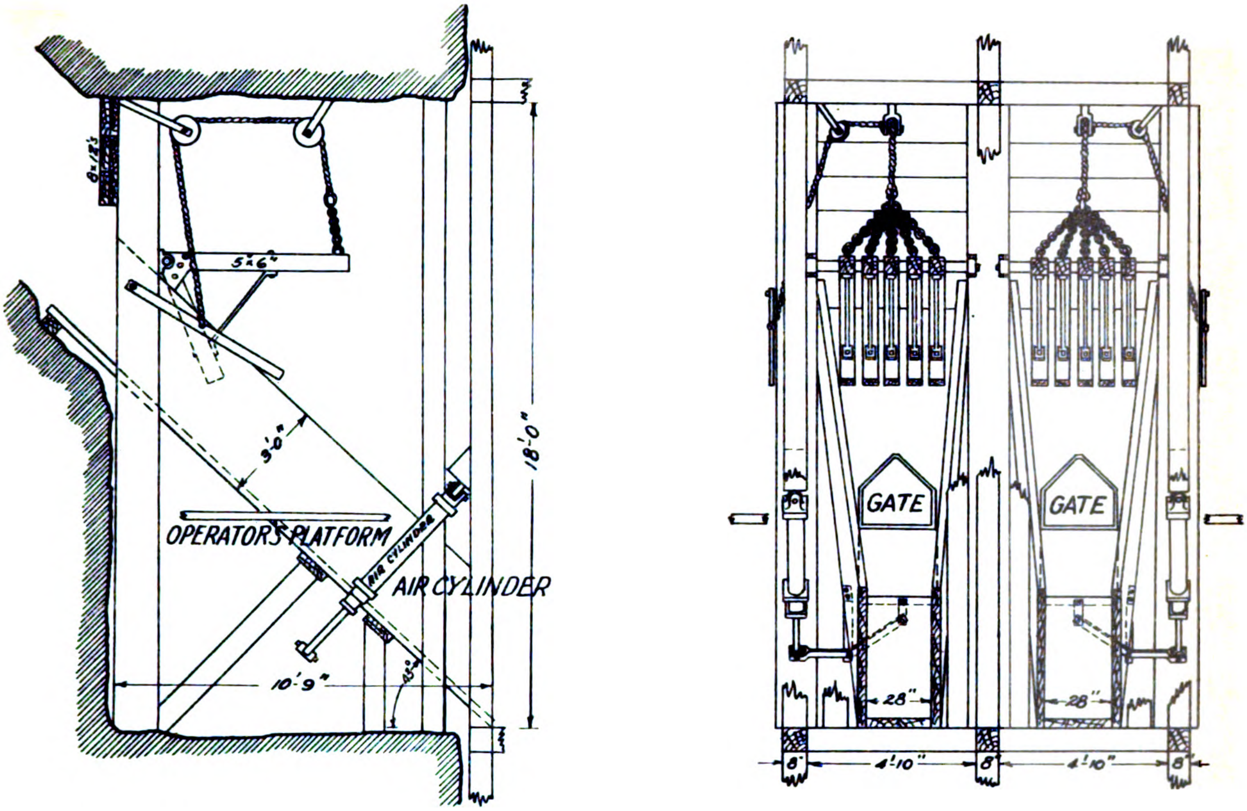


FIGURE 23.-SKIP LOADING CHUTES

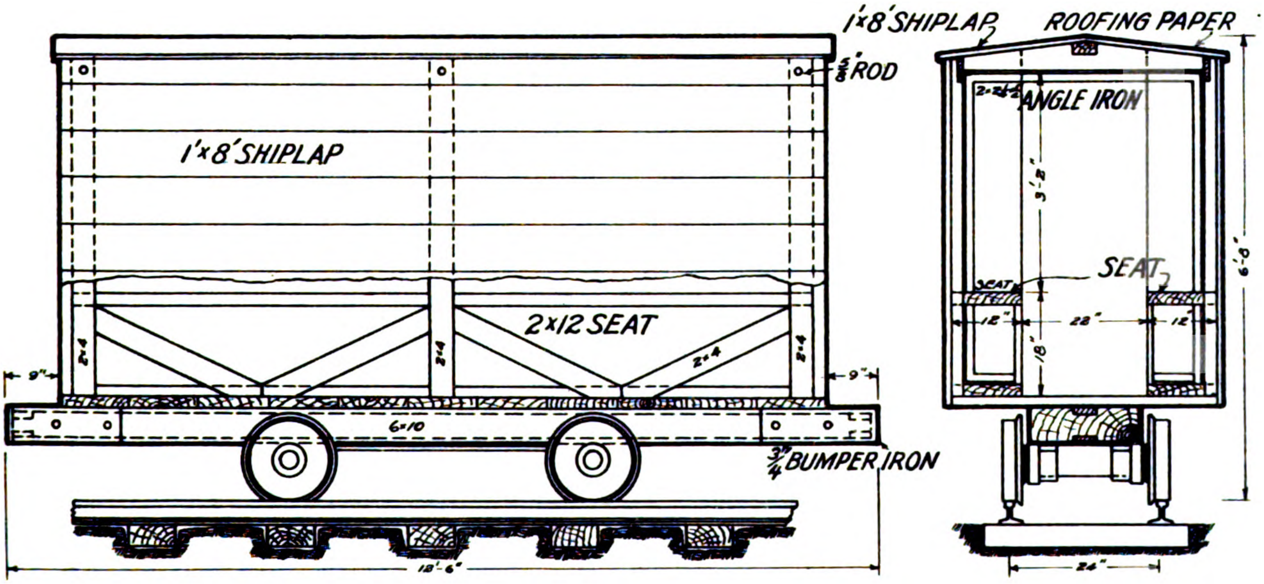


FIGURE 24.-MAN CAR